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THE COVER

Tin dredge operating in Malaysia. Malaysia is the foremost tin producing country in the world with a mine output last year of 61,400 metric tons of tin in concentrates, a little less than a third of the total free world output. Last month the country played host to the Fifth World Conference on Tin, which was held in Kuala Lumpur. The conference came at a particularly opportune time as the preceeding months had seen significant increases in the trading price of tin. *Photograph courtesy of the Malaysian Mining Corporation Group.*

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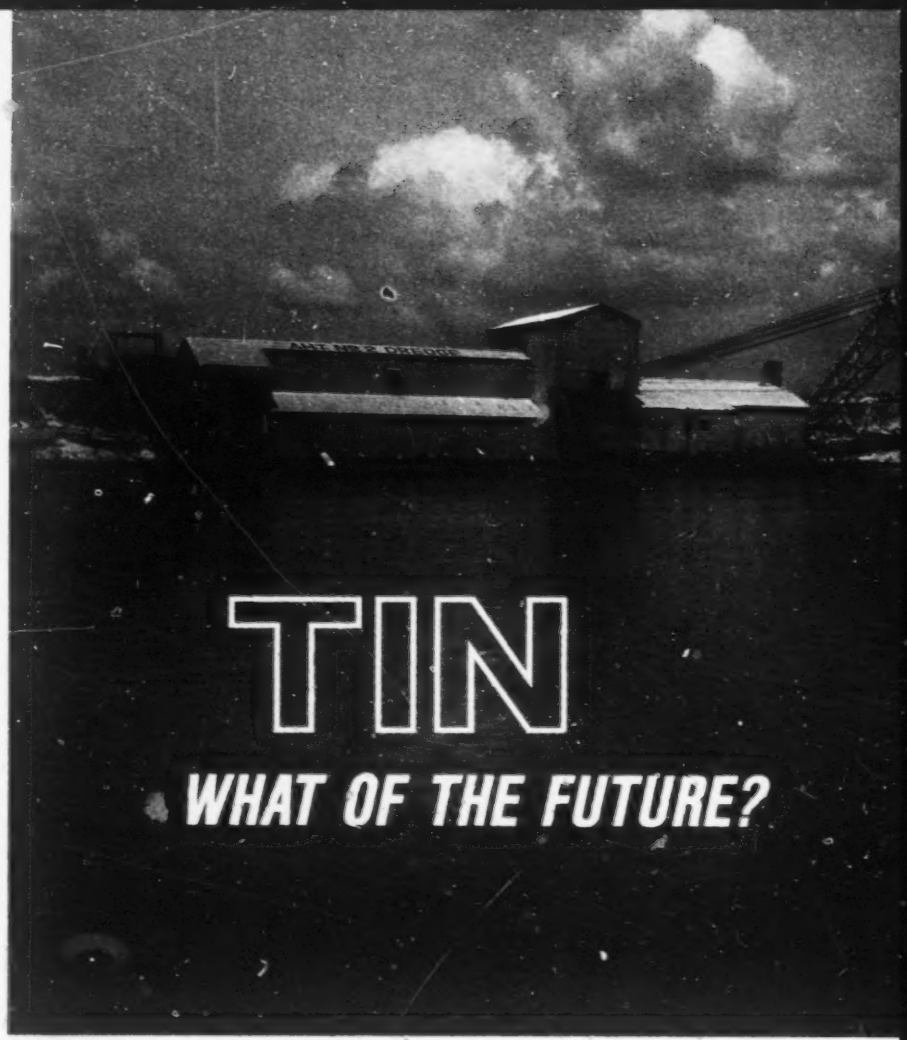


TIN

WHAT OF THE FUTURE?

At the beginning of May, the cash price for tin on the London Metal Exchange (LME) was around £5,900 per metric ton. At the beginning of July it was £6,600, since when it rose steadily, reaching £8,400 at the end of September. However, demand for the metal was low and the LME warehouse stockpile stood at around 13,000 tons, one of its highest levels ever. The international tin market was definitely intriguing and the stage was set for the Fifth International Conference.

Western world tin production in 1980, including exports from China, amounted to some 203,000 tons and the average prices per ton for the year were £7,228 or 35,700 Malaysian dollars. Current world consumption of primary tin metal is around 180,000 tons per year.

Delegates for the Fifth World Conference on Tin gathered in Kuala Lumpur, capital of Malaysia, in late October. This of course was a particularly fitting venue since Malaysia is the world's leading producing country with a mine output last year of 61,400 metric tons of tin in concentrates.

At the previous four conferences, there was little discussion about the financial and economic aspects of the tin

By **John R. Chadwick**
Associate Editor

producing and consuming industries. The broad aim of this conference was to review new production resources and consumption potential not simply in the light of the technology required for extraction or development, but also from the financial and economic viewpoints, with particular emphasis being placed on the investment that will be required to realize that potential. The intent was to cover the medium-term outlook of both the tin producing and consuming industries. These were indeed sound ideals in the light of the trends that dominated the industry at the beginning of 1981: rising production costs; a downward trend in world prices; sales from the United States strategic stockpile that posed a threat to the supply/demand situation; and problems in reaching a new International Tin Agreement. The Sixth International Tin Agreement has subsequently been drawn up and will come into force on July 1, 1982.

The conference papers were divided into four sections covering: production (new discoveries) and new production technology; consumption potential; marketing, aspects and outlook; and investment. Finally, a panel selected from among the participants discussed investment questions that arose during the conference.

The first paper, "The world tin economy and new trends" by A. La Spada, set the stage for the conference. The majority of the world's tin-bearing ores are mined in the equatorial regions of southeast Asia, Africa, and Latin America. Few of these countries are important consumers. Most tin is consumed in North America, Europe, and Japan and it is therefore widely traded between producing and consuming countries. Significant quantities of concentrates are also produced in the Soviet Union, China, Australia, and the United Kingdom. Some important figures from La Spada's paper are shown in Tables 1, 2, and

Table 1.
Production of primary tin metal

	1976	1977	1978	1979	1980
Australia	5,600	5,600	5,100	5,400	4,700
Bolivia	9,800	13,000	16,200	15,700	18,500
Indonesia	23,300	24,000	25,800	27,800	30,500
Malaysia	78,000	66,300	72,000	73,100	71,300
Nigeria	3,700	3,300	3,000	2,900	2,600
Thailand	20,300	23,100	28,900	33,100	34,700
Zaire	500	700	900	500	900
Sub-total	141,200	136,000	151,900	158,500	163,200
Percentage	77.4	75.6	78.5	78.8	81.1
Belgium/Luxembourg	4,100	3,500	3,300	2,200	2,800
Brazil	6,400	7,400	9,300	10,100	8,100
Netherlands	2,000	2,100	1,600	1,700	1,100
Singapore	—	—	1,500	4,000	4,400
Spain	5,400	5,300	4,600	4,400	4,200
United Kingdom	11,200	10,100	8,400	8,000	5,900
United States	5,700	6,700	3,900	4,500	3,900
Sub-total	34,800	35,100	32,600	34,900	31,100
Percentage	19.1	19.5	16.8	17.3	15.5
Others	6,500	8,800	9,000	7,800	6,900
Percentage	3.5	4.9	4.7	3.9	3.4
World[1]	182,500	179,900	193,500	201,200	201,200

1. Excluding mainly East Germany, China and the Soviet Union.

3. These cover production, consumption, and costs.

The second paper, "Tin production and investment" by B. C. Engel, reviewed and updated the work of the Committee on Development of the International Tin Council, which recently undertook a study on tin production and investment. The study was published in 1980 and reviewed in WORLD MINING (September 1980, page 111). In a later issue (WORLD MINING, October 1980, page 86) some very interesting production cost figures from the same report were published.

D. P. McIntyre and S. M. Richards, both of Aberfoyle Ltd., reviewed "Investment in Australia's future tin resources." They stated that hard rock reserves will continue to be the source for most tin output over the next ten years. Research and development will concentrate on improving recovery through preconcentration techniques, flotation, and possibly fuming. Constraints on new mine developments are the exchange rate of the Australian dollar and the costs of labor, energy, and construction. These factors favor the expansion of existing mines and developments in their vicinity rather than the development of new mining areas. Any new developments are likely to be open pit mines at grades of 0.5 percent Sn or higher. Favorable metallurgical characteristics will also be an important consideration.

Tin production in Brazil has been increasing significantly over the last five years as a result of major exploration investment, mainly in the Amazon region. "New tin discoveries in Brazil" were reviewed by J. R. M. Franco. A major deposit containing 23,000 tons of tin has been discovered on the Pitinga River on the north bank of the Amazon basin, 300 kilometers (186 miles) north of Manaus. Surucucus is another important recent discovery. This tin deposit is situated to the west of Roraima Federal Territory in northern Brazil. Garimpeiros (pick-and-shovel miners) had produced 563 tons of tin concentrate over a 14-month period. However, mining has been suspended for the time being because of troubles between the miners and local Indians.

Goiás Mineracão is carrying out an intensive exploration program in the Serra Dourada. The company expects to invest $4,000,000 in the next three years to evaluate an extensive tin ore body. The Paranapanema group has invested some $3,700,000 to develop the important São Raimundo deposit in the southern central Pará State, on the Xingú River. Exploration results to date have shown an alluvial deposit with reserves of 3,407 tons of tin, at a grade of 0.408 kilograms Sn per cubic meter (0.688 pounds Sn per cubic yard). Open pit mining is proceeding with a production rate of some 3 tons of cassiterite a day.

The major discoveries over the last 10 years in Indonesia have been offshore reserves, greatly benefiting this country's output. Important deposits have been discovered in the Kundur Sea. Onshore, the discovery of the Tempilang deposit is regarded as significant since it is of a type to which very little attention has been paid hitherto. It is hoped that development of both sources will begin soon. In their paper, "Review of discoveries of new tin deposits in Indonesia," S. Sujitno and M. Simatupang describe the geology of these deposits and review current exploration work.

New discoveries of deposits and occurences, both onshore and offshore, including geochemical anomalies, were summarized by S. S. Rajah and D. S. Singh in "Discoveries of tin deposits and occurrences in Peninsular Malaysia during 1974 to 79." The present status and future prospects of the industry were presented as well as recommendations for exploration targets.

"Possible future tin mining activities and possible new mines in Nigeria" were covered by I. Gombe. Alluvial sources have, hitherto, provided all the cassiter-

Table 2.
Consumption of primary tin metal

	1976	1977	1978	1979	1980
Australia	3,600	3,800	3,600	3,400	3,000
Belgium/Luxembourg	3,000	3,300	3,100	2,400	2,700
Canada	4,800	5,300	4,900	4,700	4,800
Czechoslovakia	3,500	3,100	3,200	3,200	3,200
France	10,200	10,700	9,900	9,700	10,100
West Germany	14,800	14,100	13,500	13,700	14,300
India	3,000	2,700	2,600	2,500	2,500
Italy	5,900	6,200	5,800	6,000	6,500
Japan	34,700	29,700	29,600	31,200	31,200
Netherlands	3,800	4,900	4,800	4,800	5,000
Poland	5,100	4,700	4,600	3,900	3,300
Romania	3,100	3,200	3,200	3,200	3,300
Spain	4,600	3,700	4,500	4,100	4,300
United Kingdom	13,500	12,700	12,200	11,100	7,800
United States	51,800	47,600	48,400	49,500	46,000
Sub-total	165,400	155,700	153,900	153,400	148,000
Percentage	85.3	84.3	83.6	83.7	83.4
Others	28,600	29,100	30,100	29,900	29,400
Percentage	14.7	15.7	16.4	16.3	16.6
World[1]	194,000	184,800	184,000	183,300	177,400

1. Excluding mainly East Germany, China and the Soviet Union.

Table 3.
Weighted average of tin production costs and prices in Malaysian dollars per pikul

	1976	1977	1978	1979	1980[2]
Average cost[1] yearly	1,000	1,264	1,408	1,534	1,824
Index (1971 = 100)	179	227	252	275	327
Average tin price (Penang) yearly	1,147	1,588	1,743	1,961	2,160
Index (1971 = 100)	181	251	276	310	342

1. Including royalties, export duties, and tributes. 2. Provisional figures.

ite mined in Nigeria, but annual production over the last decade has fallen from 11,200 to below 4,000 tons. The major reasons for the decline are the depletion of easily mined high value alluvial reserves, the slow introduction of mechanization, increased operating costs, and the absence of alternative reserves.

Primary deposits offer good prospects to expand reserves and the Nigerian Mining Corporation is actively exploring such possibilities. Additionally, there are large known reserves of subbasalt cassiterite and the prospect for further such reserves is high. These two sources offer great potential for production expansions. Similarly future exploration is expected to increase alluvial reserves and activity in that sector.

S. Kulvanich's paper, "Offshore exploration for tin in Thailand," reviewed an exploration program for tin and heavy minerals in deep water, 30 to 45 meters (98 to 148 feet), in the Andaman Sea, southwest of Thailand. The first phase of the project involved the compilation of geological data and a geophysical investigation, and gave promising results. A supplemental drilling program is now underway to determine the distribution, extent, and characteristics of the prospects.

Zaire is another country where tin output declined dramatically, to below 4,000 tons a year, over the last decade. A paper prepared by that country's Department of Mines outlined "Tin in Zaire and its prospects for the future." The government has produced a recovery plan aimed at improving production and conditions, increasing research and exploration, and optimizing recovery profitability. Sominki, operating in the Kivu-Maniema region and Zairetain, operating in Shaba province, are the main producers. A new company, SMDG, has started mining small alluvial deposits. Société Minière de Katondo and Société Minière de Kania will commence tin mining in the future, while Enterprises Minières Zairoises (Emza) is to restart tin mining operations.

"The Kuala Langat tin field" was examined by Yap Cheng Hock. The area already includes active mines in the Dengkil area of Selangor but the development of the deep ore body in Brooklands Estate and the Forest Reserves to the south will increase Kuala Langat's importance in the future. The paper outlined exploration undertaken between 1973 and 1975 and the problems encountered. Multiple mineralization sources, from the granite contact zones at Ayer Hitam in the north and Sungei Besi and Kajang in the northeast, are suggested as well as at least three phases of mineralization.

The South Selangor, or Kuala Langat, deposit is the world's largest known tin reserve, estimated to contain more than 200,000 tons of tin. It is covered with overburden varying in depth from 20 to 30 meters (60 to 100 feet). The ore body itself varies in depth between 50 and 100 meters (150 to 300 feet). This great depth requires "An alternative method for mining deep alluvial ore deposits," as described by J. M. Donkers.

The new concept employs a suction dredge, equipped with a dredging wheel capable of recovering paydirt from a depth of 100 meters (300 feet), and a bucket line dredge for removing the overburden. The combined throughput would be in excess of 19,000,000 cubic meters (25,000,000 cubic yards) per year. Donkers' discussed the proposed equipment, a possible mine plan, and capital and operating costs. The possibilities and limitations of a variation of the dredging wheel dredge for offshore mining applications were also covered.

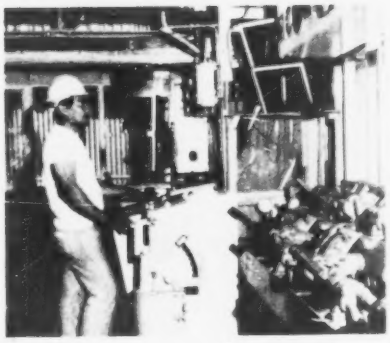
Continuing with dredging equipment was A. W. K. McDowell's paper "A review of 16 years' development of the underwater bucket wheel excavator and its current application to alluvial tin mining." The development and evolution of this equipment was detailed, pointing out the problems encountered and their solution, and the future potential for such units. Results achieved at two operations in Brazil were described in detail.

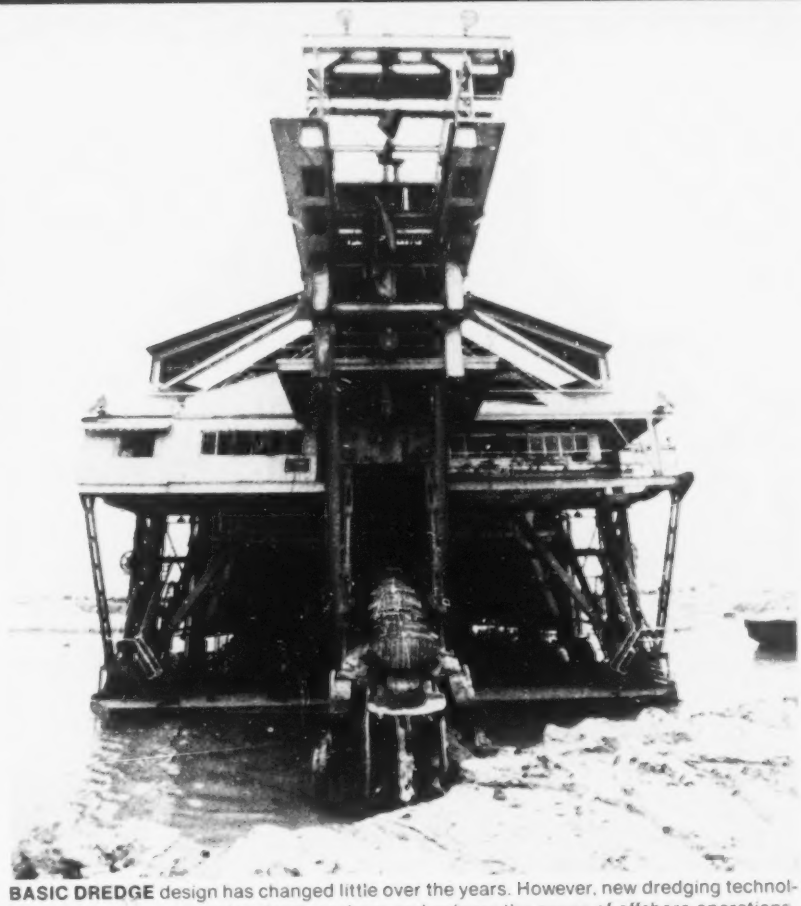
"Tin dredging operations at Berjuntai Tin Dredging Berhad, Selangor, West Malaysia" were presented by Lim Che Wan. This is one of the largest alluvial operations in Malaysia, with a current output of over 300 tons of concentrate per month. Current developments include the building of a new dredge and the modification of two others. One of the two dredges to be modified will be equipped with an experimental underwater bucket wheel.

Looking to a different, though still traditionally Malaysian, mining method, Hew See Tong examined "Investment in, and operation of, a gravel pump mine in typical dredged out land today." Over the last ten years there have been considerable changes in both investment requirements and mining methods for gravel pump mines. These have mainly been as a result of the need to operate in lower grade ground and rising operating costs. Typical gravel pump mines employ conventional open pit mine stripping techniques for overburden and monitoring for ore mining. As a result, capital requirements have increased vastly with earth moving equipment accounting for a high proportion of this capital.

"A regional approach to research on tin production" by A. H. Hassan dealt with problems faced by the tin mining industry in Indonesia, Malaysia, and Thailand, and their possible solutions. Many of the problems are common to all three countries because of similarities in deposits and extraction methods. Recog-

GRAVEL PUMP mining is being particularly hard hit by increasing production costs.





BASIC DREDGE design has changed little over the years. However, new dredging technology may increase operating depth onshore and enlarge the scope of offshore operations.

nizing this, the countries have formed the South East Asia Tin Research and Development Centre (SEATRAD).

Current research projects include the development of techniques for the exploration and mining of primary tin deposits, reducing costs by improving hydraulic mining techniques, development of drilling and sampling techniques for detrital tin deposits, and process control of treatment plant operations.

Turning specifically to the treatment side of the operation, W. P. Cross presented "Development of molded rubber plates for use in trommel screens." The current use of such screen plates for undersized material, after being discharged from the dredge buckets into the trommel screen, is the first time this locally-produced material has successfully been used in such applications since Peninsular Malaysian tin dredging commenced. Cross traced the development of a design which requires no reinforcing material and utilizes a clip-in principle.

The advantages claimed for molded rubber screen plates include their nonblinding properties, longer life than the conventional high carbon steel screen plates used previously, less maintenance, and ease of replacement. The application of such screens is on the increase.

The paper entitled "World tin consumption—present position and medium-term forecast," by U. Yahya and N. C. P. Hollands, moved the conference on to the second section covering consumption. Other papers in this section were: "Tinplate;" "Tin conservation in the canning industry;" "Solder;" "Canning;" "Tinplate can consumption potential in Asean;" "Material substitution and the demand for tin;" "Tin consumption meets the challenge from technological change;" "Recycling of packaging materials;" and "Consumption research benefits the tin industry."

J. C. Bowi opened the third section (Marketing—aspects and outlook) with a paper that considered the imbalance between physical tin supplies channelled through Penang and London against those marketed direct to consumers. "Tin marketing—polarization of interests between the producing and consuming element of the international tin industry." The four other papers in this section covered: "Market quotes and marketing practices;" "Establishment of a tin exchange in Malaysia;" "The use of the London Metal Exchange for hedging purposes;" and "The role of tin within the U. S. national defence stockpile."

Six papers were presented in the final section on investment, four covered production investment while two covered consumption investment. The first, by J. H. Forsyth, was "Problems affecting the financing of long-term development projects."

A paper prepared by members of Billiton International Metals BV examined "Considerations relating to investments in tin mining." The paper related particularly to offshore mining. The trend in tin mining is increasingly towards capital intensive, large-scale operations which, by their very nature, entail relatively high fixed costs and relatively low variable costs. For capital intensive investments, the rate at which the investment can be earned back is as important as the final profits, while for gravel pump mining—with relatively low capital investment—the governing criterion is more likely to be the profit attained.

The two remaining papers covering production investment were "An alternative method for projecting the evolution of the tin economy, with particular emphasis on investment," by H. Hashimoto, and "Some problems anticipated in developing a deep-seated alluvial tin deposit in Malaysia," by Thong Kean Sin. The latter covered a subject that will become a necessity in the near future if Malaysia's tin production is not to decline too sharply. The paper deals with the problems anticipated in mining a deep-seated alluvial deposit consisting of at least 76 meters (250 feet) of alluvium overlying soft mudstone. A possible mining method may be to operate a dredge in a deeply depressed paddock, employing pre-stripping of overburden ahead of the dredge.

The final two papers were, "Consumption investment in Japan" by K. Tanaka and "The tinplate industry in the European Economic Community—aspects of its future development." For further information about the conference and the papers, write to The Secretary, The International Tin Council, Haymarket House, 1 Oxendon Street, London SW1Y 4EQ, England, or to The Secretary, Steering Committee for the Fifth World Conference on Tin, Ministry of Primary Industries, Gurney Road, Kuala Lumpur, Malaysia. ■

The conventional sinter plant-blast furnace process currently accounts for 86 percent of all lead smelted in the western world. Economics of such a plant are examined through a western Canada feasibility project

Evaluating the economics of lead smelting

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Department of Energy, Mines, and Resources
Ottawa, Canada and
George M. Meisel
Consulting Metallurgical Engineer
El Cerrito, California, United States

Mining companies with integrated smelting and refining activities tend to show greater stability and enjoy greater flexibility and control in marketing than do firms engaged only in mining. Integrated operations may also offer benefits through higher metal recoveries in mining and smelting and, in certain exceptional cases, an integrated approach may even be necessary to achieve acceptable overall metal recoveries. This article is intended as an example of how to evaluate a lead smelting and refining operation. From this an evaluation of future lead smelting and refining opportunities in Canada is presented. Capital and operating costs and revenues have been updated from a previous evaluation of this project, conducted in the fourth quarter of 1978.[1]

The smelting and refining operation is based on production of 100,000 metric tons of lead per year. The analysis is based on a site on the Pacific Ocean coast of British Columbia.

The concepts for this initial evaluation of the lead smelting and refining complex are based on the following criteria:

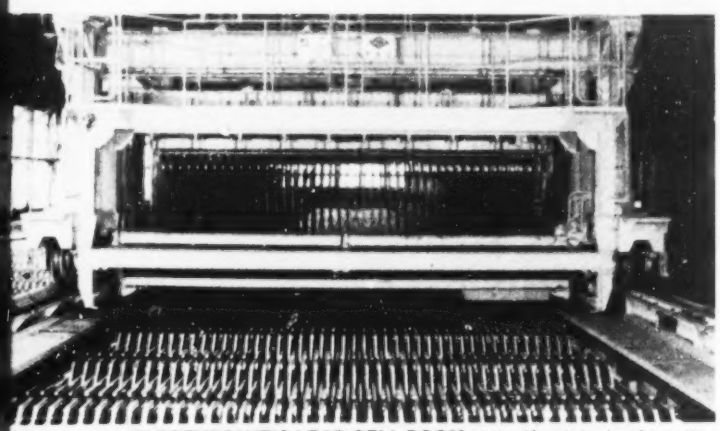
- Compatibility with the requirements for processing lead concentrates of an analysis expected in western Canada
- Meeting the constraints imposed by the environment and the logistics of the area
- Availability of capital and operating cost data from existing operations sufficiently similar to permit factoring for the proposed complex.

Best practicable current technology, for which reasonably reliable capital and operating costs are available, is the basis for the economic evaluation. It is acknowledged that current technology has limitations in terms of environmental compliance, although careful engineering has tranformed both sintering and blast furnace smelting from dirty operations to ones which, at a price, can approach the constraints of present environmental, safety, and health regulations. A greater problem is that of preventing lead from entering the plant environment from a fire refining operation. For this reason and the availability of relatively inexpensive hydro-electric power, an electrorefining operation is included in this study instead of fire refining.

The operation conceived for this study is based upon well-proven technology, i.e., the conventional sinter plant/blast furnace process, which still accounts for 78 percent of the total number of plants and 86 percent of lead capacity in the western world,[2] plus electrorefining which is commercially well established in Japan and likely to supplant fire refining in new greenfield plants.[3] Concentrates are sintered, and the sinter reduced in a blast furnace. Lead bullion tapped from the blast furnace is cast into anodes and refined electrolytically.

A sulphuric acid plant with a bleaching stage is included to recover the sulphur dioxide from the sinter off-gas.

Order-of-magnitude capital and operating costs and revenues are estimated on this basis indicating that further investigation appears justified. Costs are estimated from data gathered from lead concentrate sintering and blast furnace smelting plants built in Missouri before 1970 and from two electrolytic refining circuits built in Japan in 1969 and 1975. P. J. Mackey[4] has referred to a smelter built in India in 1978, but the authors are unaware of any other world-scale primary smelters built since 1970. Costs of the Indian plant, even if available at the time of writing, would have been of limited use for a western Canada plant. The results reported here are as yet based on a generalized location on the Pacific Coast in the Kitimat/Prince Rupert area of British Columbia.



ELECTYROLYTIC LEAD CELL ROOM at the Kamioka lead smelter of Mitsui Mining and Smelting Company Limited, Gifu Ken, Japan.

New Process Technology

While the study is based on conventional technology, some mention should be made of improved technology which may become incorporated in future plants. New lead processes under development include several based upon direct pyrometallurgical reduction of concentrates and some on chloride leaching. The direct reduction processes all have in common the elimination of sintering, although from the point of view of gas and dust containment, this advantage is lost to some extent in certain of the processes in which bullion and/or slag must be transferred to and treated in further stages.

Boliden Lead Process: In the Boliden lead process, which represents an early attempt at direct reduction and which has been commercially operated since 1963, dried concentrate and fluxes are fed into an electric furnace through the roof, producing a bullion containing 3 percent sulphur and a slag containing 4 percent lead. The lead is tapped periodically into one of two converters and blown to burn off the remaining sulphur. The high-lead slag is tapped into ladles from which it is poured into a slag fuming plant. Overall dust carry over in off-gases is reportedly about 42 percent of concentrate feed.[5] Refining is

carried out in kettles. The process has never been adopted by other companies.

The consumption of energy in the Boliden direct reduction process is reported to be significantly less than traditional processes.

Top Blown Rotary Converter Process: The environmental disadvantage of having to transfer high sulphur bullion to a separate converting stage has led Boliden to adapt the top blown rotary converter to the smelting of lead concentrate with all stages carried out in one vessel.[7] As a result of this development, Boliden is planning a top blown rotary converter plant for the treatment of about 100,000 tons of high grade concentrate per year. This suggests that Boliden may be on the point of replacing its electrothermic direct reduction process. In the proposed new plant, oxygen enrichment of blast air will reduce off-gas volumes and, as with other top blown rotary converters, it is supposed that dusty off-gases will be contained by totally enclosing the furnace and tapping area, which will be vented to dust collection.

Three other examples of new lead extraction processes have been developed to various stages and may become available for commercial application within a few years. These processes appear to offer prospects for combining good metallurgical performance with comparatively inexpensive environmental control, i.e.:

Kivcet-CS Direct-Lead-Zinc Smelting Process: The Kivcet-CS direct lead smelting process is being developed under license from Russia by Klöckner-Humboldt-Deutz-Wedag (KHD). The trademark is the Russian acronym for "oxygen-flash smelting-cyclone-electrothermics," and CS stands for lead-zinc. However, a shaft furnace is used for the oxygen flash smelting of lead-zinc concentrates, different from the cyclone furnace used in the older and better known Kivcet-LZ process for copper-zinc concentrates.[8] Elimination of both the sintering and converting stages is achieved in this process. The concentrate is flash roasted and smelted in suspension using oxygen and the molten metal/metal oxide mixture then enters an electric furnace section adjoining the flash chamber, where lead and zinc oxides are reduced with coke. From results of pilot plant tests by the UNIICVETMET Institute at Ust-Kamenogorsk, Russia, KHD has designed industrial scale installations,[9] and is about to construct one in Bolivia.

QSL Direct Lead Smelting Process: The Queneau-Schuhmann-Lurgi process for direct smelting of lead concentrate is sponsored by St. Joe Minerals Corporation, Berzelius Metallhutten GmbH, and Lurgi Chemie & Huttentechnik GmbH. The principal economic and environmental benefits of this process are obtained by eliminating the sintering step. Fluxed concentrate pellets, powdered coal, and oxygen are fed to the reactor, and bullion, slag, and high strength SO_2 flue gas are discharged. The physical shape of the reactor is a cross between an SL/RN direct iron ore reduction kiln and a copper converter. This process has been developed to a large scale batch demonstration stage at Duisburg, West Germany. The results have been published in abstract form.[10]

Ferric Chloride Leach-Fused Salt Electrolysis Process: A ferric chloride leach and fused salt electrolysis process has been developed to a large scale continuous pilot stage at the U. S. Bureau of Mines Metallurgy Research Station, Reno, Nevada. The work is being sponsored as a joint government-industry effort by St. Joe Minerals Corporation, Amax Lead & Zinc Inc., and others. The encouraging results of prior stages of this work have been published by the U. S. Bureau of Mines.[11]

Imperial Smelting Process: A further alternative would involve the treatment of bulk or mixed lead/zinc concentrates by the Imperial Smelting Process (ISP). This process still requires sintering of the concentrates, but is in some ways a less far-reaching predecessor of the Kivcet-CS process. Given that Canadian lead concentrates are invariably prepared from lead/zinc ores, the ISP offers the potential for enhancing integrated mill/smelter recoveries further. The ISP should always be considered in the Canadian context because of its potential benefits in recoveries and shared infrastructure costs in an integrated operation.

Assumed Plant Criteria

In this exploratory review of plant economics there is no attempt at site comparison. The model is based on the Kitimat/Prince Rupert region at navigable tidewater, while recognizing that other physically suitable locations may exist in western Canada. Concentrates could be delivered by barge (as assumed here) or by rail from existing or future mines in British Columbia, the Yukon, or the Northwest Territories.

The size of any new primary smelter must satisfy certain basic requisites. It must be assured an adequate supply of concentrate and other operating materials, and be able to dispose of its metal products and byproducts, notably sulphuric acid, but is must also be large enough to achieve reasonable economies of scale. Potential concentrate supply may be enough to justify a much larger plant but it is thought that a nominally "minimum" world scale primary smelter of 100,000 tons capacity may be all that the markets might initially bear. However, the plant design must provide for later addition of a second blast furnace.

No attempt is made here to define the source(s) of concentrate for a western lead smelter. At present, the main sources of

Schematic materials balance in tons per year

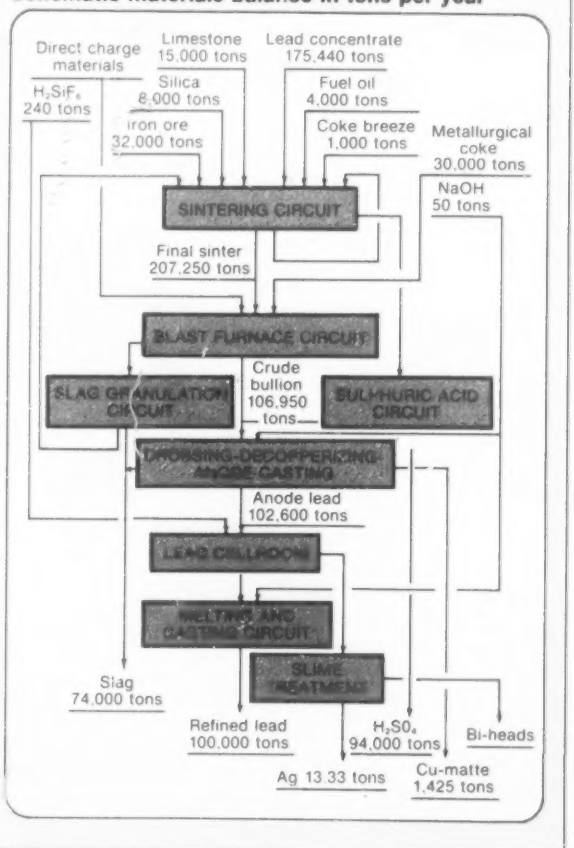


Table 1.

Content	Analysis	Recovery, percent
Lead	60 percent	95
Zinc	8 percent	0
Iron	7 percent	0
Silver	80 grams per metric ton	95
Copper	0.5 percent	65
Sulphur	18 percent	97

Table 2.
Order-of-magnitude capital costs

	Dollars
Receiving, proportioning, sintering	22,800,000
Blast furnace, slag granulation	11,300,000
Refining, melting, casting, shipping	26,000,000
Sulphuric acid (including bleaching stage)	16,200,000
Barge dock	5,100,000
General plant, yard, office facilities	12,000,000
Liquid effluent control	5,800,000
Subtotal	99,200,000
Contingency (20 percent)	19,800,000
Total constructed costs	119,000,000
Plus:	
Local sales and use taxes (waived)	—
Land acquisition (500 acres, nominal)	1,200,000
Mobile plant equipment	1,500,000
Personnel training and plant start-up	1,500,000
Spare parts inventory (3 percent of equipment)	1,900,000
Owner's administrative cost	3,700,000
Working capital from Table 3	16,200,000
Total other costs	26,000,000
Total capital costs	145,000,000

Table 3.
Working capital requirements

Item		Months required	Dollars
Operating materials and supplies	(from Table 6)	3	616,000
Fuel oil	(from Table 6)	3	531,000
Coke	(from Table 6)	3	1,335,000
Power	(from Table 6)	1	39,000
Maintenance materials	(from Table 6)	3	477,000
Payroll	(from Tables 4, 5)	1	689,000
Concentrates in storage and process	(from Table 9)	2	12,468,000
Total			16,155,000

Table 4.
Salaried staff and annual costs

Job	Number	Annual costs, dollars
Smelter manager	1	55,000
Production manager	1	43,000
Administration manager	1	34,000
Maintenance manager	1	43,000
Chief metallurgist	1	38,000
Chief engineer	1	38,000
Comptroller	1	38,000
Purchasing agent	1	34,000
Superintendant environmental safety and health	1	38,000
Superintendant quality control	1	38,000
Superintendant maintenance shop	1	38,000
Superintendant electrical and mechanical	2	76,000
Department superintendents	4	152,000
Security officer	1	26,000
Head nurse	1	24,000
Shift foreman	14	392,000
Maintenance foremen	4	112,000
Production engineers	3	90,000
Buyer, warehousemen, accountants	9	198,000
Plant engineers	3	90,000
Personnel administrators, miscellaneous	3	72,000
Analysts	4	96,000
Guard and scale house	8	160,000
Salaried subtotal	69	1,925,000
Burden 35 percent		674,000
		2,599,000

exported lead concentrate from western Canada are Cyprus Anvil Mining Corporation in the Yukon Territory and Pine Point Mines Ltd. in the Northwest Territories, both of which are presently tied to export sales contracts while Pine Point also supplies the Cominco plant at Trail, B.C. Over the longer term, other ore deposits will be developed in western Canada and adequate quantities of concentrate will become available.

The schematic materials balance shown in the figure is based on the assumption that the average analysis of the lead concentrates and the recoveries of their values contained are as shown in Table 1.

Dock and storage facilities are provided for both receiving concentrate and shipping sulphuric acid by barge. Barge delivery of concentrate is assumed not to require thawing facilities. However, thawing would be necessary if concentrate were transported by rail. Metal production is assumed to be shipped from the plant by truck, rail, or sea using existing facilities.

The charge proportioning operation for the sinter strand is assumed to be fully mechanized with weigh feeders equipped with modulating controls. Sintering utilizes an up-blast system with recirculation of weak sulphur dioxide gas from the discharge end of the sinter strand.

The hot zone of the blast furnace is water jacketed. A Roy tapper provides a continuous discharge of bullion and slag to a settler for separation.

Slag discharged from the settler is granulated and laundered to a sump from which it is transferred by bucket elevator to a load-out and haulage to the slag dump. A slag fuming furnace for zinc recovery is not considered in the present model. An electrothermic unit could be added for this purpose whenever the zinc market justifies the high power and coke requirements per ton of zinc recovered, i.e., 7,200 kilowatts per hour and 390 kilograms coke per ton of recovered zinc.[4]

Crude bullion is transferred from the settler to drossing kettles by ladle. Copper is recovered as matte from a small electrothermic or reverberatory dross furnace; the de-copperized bullion is cast into anodes for electrolytic refining. It is assumed that the matte contains 40 percent copper and 35 percent lead.

Electrolytic refining is carried out in a cell house equipped with automated "Jumbo" anode and cathode handling, and forced draft ventilation, see photograph on page 48.

Dust and fume recovery points include the charge proportioning and charging, sintering and sinter handling, blast furnace charging and blast furnace tapper-settler, bullion drossing, and de-copperizing, anode casting, refined lead melting and casting, and other dry handling plant sections.

For control of liquid effluent, a two-stage lime precipitation and settling circuit is included for heavy metals removal and plant water reclamation from all gas scrubber bleeds and other effluent sources. In the present exploratory economic study no attempt is made to anticipate specific environmental regulations.

Evaporative cooling of off-gases in the dust collection train is accompanied by injection of pulverized limestone to prevent bag house fires which may result from pyrophoric dust. Dust from the bag house is returned to the sinter strand.

The acid plant, with a capacity of approximately 94,000 tons of sulphuric acid per year, employs two-stage conversion and absorption. A bleaching stage is included to make the otherwise "black" acid more easily marketable.

Capital Cost Estimate

The capital cost estimate summarized in Table 2 is expressed in fourth quarter 1980 Canadian dollars; it is non-specific as to site, and is prepared by factoring and escalating

the costs of existing respective plant circuits in Missouri and Japan. Thus the estimate is of order-of-magnitude, and a 20 percent allowance has been provided for contingencies as well. The costs of engineering and procurement have been pro-rated for each plant section. Originally prepared in U. S. dollars and yen, the estimates have been updated and adjusted to Canadian dollars and reflect current cost conditions in western Canada.

It is assumed that 36 months will be required for construction of the plant, followed by six months for tune-up to design capacity.

The allowance made for personnel training and plant start-up costs amounts to about one month's direct operating costs. The owner's administrative cost allowance, before and during construction, is based on actual Canadian plant experience, although it can vary greatly.

Operating Cost Estimate

Operating costs in fourth quarter 1980 Canadian dollars are developed by factoring the requirements for labor and the quantities of supplies and utilities for existing plants in Missouri and Japan. Costs are assigned to these quantities directly in Canadian dollars using unit costs applicable to the Kitimat area.

Total employment is 275, including 69 salaried and 206 hourly paid employees. Table 4 shows the distribution of salaried staff, salaries, and 35 percent allowance for payroll burdens and fringe benefits.

The distribution of hourly paid personnel and annual costs for each plant section is shown in Table 5. Estimated average wage costs of $9.42 per hour for smelter workers and $10.48 for maintenance workers are based on 40 hours per week and 52 weeks per year: this amounts to $19,600 and $21,800 per man-year respectively. After adding a further 7.5 percent overtime allowance and 25 percent for payroll burdens and fringe benefits, the total annual costs per person become $26,338 and $29,294.

Annual quantities of operating supplies and utilities, together with unit costs, and total annual costs for 100 percent plant capacity, are shown in Table 6. Where applicable, the unit costs include freight to the Kitimat area.

Table 7 is a summary of all operating costs, recapitulating the rounded-off totals of operating supplies and utilities, labor costs, maintenance supplies, plus allowances for property tax and insurance, all in constant end-of-1980 dollars. Costs are also shown for the first year of production, in which it is assumed that 95 percent capacity is achieved by the end of six months giving an average first year production of 72.5 percent, and for subsequent years during which an average 95 percent utilization is assumed.

Table 5.
Hourly paid personnel

Labor rates	Dollars	
	Operators	Maintenance
Average hourly wage rates	9.42	10.48
Average man-year wage (2,080 hours)	19,600	21,800
Overtime (7.5 percent)	1,470	1,635
Subtotal	21,070	23,435
Burden and fringes 25 percent	5,268	5,859
	26,338	29,294

Manning distribution Plant sections	Persons	Annual costs (dollars)
Materials handling, sinter, acid	40	
Blast furnace, slag handling	24	
Dross, de-copper, de-tinning	16	
Cellroom, melt, cast, ship	28	
Yard, change house, etc.	16	
Subtotal operating labor	124	3,266,000
Maintenance and shop labor	82	2,402,000
Total	206	5,668,000

Return on Investment

In this conceptual stage, where the feasibility of an additional lead smelter in western Canada is being analyzed, it is intended to present only a general indication of its rate of return. More detailed analysis will be necessary when more

Table 6.
Operating supplies and utilities (at 100 percent capacity)

Item	Kilogram per metric ton of lead	Annual quantity (metric ton per year)	Unit cost (delivered) (dollars per metric ton)	Annual costs (dollars)
Sinter charge flux:				
Iron ore	320	32,000	35	1,120,000
Silica	80	8,000	12	96,000
Limestone	150	15,000	30	450,000
Subtotal				1,666,000
Blast furnace flue treatment:				
Limestone fines	10	1,000	30	30,000
Drossing:				
Soda ash	16	1,600	248	397,000
Sulphur	10	1,000	100	100,000
Subtotal				497,000
Anode and pig cast:				
Caustic soda	0.5	50	840	42,000
Refining:				
H_2SiF_6 (25 percent)	2.4	240	175	42,000
Glue	0.45	45	2,310	104,000
Subtotal				146,000
Liquid effluent control:				
Lime	5	500	165	83,000
Total operating supplies				2,464,000
Fuel oil:				
Sinter (high S)	40	4,000		
Blast furnace	80	8,000		
Anode and pig cast	12.8	1,280		
Subtotal, fuel oil		13,280	1,690	2,125,000
Coke:				
Lump, blast furnace	300	30,000	175	5,250,000
Breeze, sinter strand	10	1,000	90	90,000
	kilowatt hour per metric ton	megawatt hours	cents per kilowatt hour	
Power:				
Dross furnace	85	8,500	1.64	
Refining	125	12,500	1.64	
Services	75	7,500	1.64	
Subtotal power	285	28,500		467,000
Total, fuel and power				7,932,000
Maintenance supplies (3 percent of equipment and piping)				1,909,000
Total operating supplies				12,305,000

Table 7.
Operating costs at various plant operating rates*

	Annual costs, percent dollars	
	100 percent capacity	95 percent capacity
Operating materials and supplies	2,464,000	2,341,000
Fuel and power	7,932,000	7,535,000
Maintenance supplies	1,909,000	1,909,000
Salaries, wages, benefits	8,267,000	8,267,000
Property tax, insurance (1.5 percent of constructed cost)	1,785,000	1,785,000
Total operating costs	22,357,000	21,837,000

*Note: Employment costs, taxes, and insurance are not pro-rated.

precise information is available on capital and operating costs at a specific site, on corporate structure, and on general, administrative, and sales expenses.

Smelter revenues are estimated on an average of smelter schedules applying in the European Common Market and Japan. The proposed smelter will compete with the existing lead smelters in those two areas for lead concentrates from western Canada. Generally, smelter schedules provide for payment for the recoverable values in the concentrates at prevailing price levels, reduced by certain deductions, treatment charges, and penalties for impurities in troublesome quantities. The smelter schedules generally include formulae tying the treatment charges to the price of lead and/or the cost of energy and labor. While details of individual deductions and/or treatment charges vary from one smelter schedule to another, there is little difference in a normal competitive market in the net return to the smelter or to the mine from which it buys the concentrates.

Plant revenues are based on an average 95 percent of capacity. Metal prices have been highly volatile during the past two years. However, the net effect on smelter revenues from the increased silver price is outweighed by the reduced lead price. The following net prices, freight-on-board smelter in Canadian dollars, were used in estimating revenues, as of end October 1980:

Lead:	$(Canada)0.53 per pound, corresponding to $(U.S.)0.45.
Silver:	$(Canada)22.29 per troy ounce, corresponding to $(U.S.)18.95.
Copper:	$(Canada)0.67 per pound contained in matte, reduced for the cost of converting and refining at $(U.S.)0.95 per pound cathode copper.
Sulphuric Acid:	$(Canada)10.00 per ton.

Results of the original study presented in April 1979 show that the return on the original investment would improve approximately 2.3 percent, if the proposed smelter were to become part of an integrated mining and smelting operation; such an integrated operation would benefit from federal and provincial tax provisions permitting part of smelter construction costs to be written off against mining profits.

A sensitivity analysis conducted as part of the April 1979 study indicates the approximate effect on the return on the original investment of variations in the lead price, treatment charges, energy cost, capital cost, and operating rate, as shown in Table 10.

Table 8.
Estimated return on original investment (based on end October 1980 costs and revenues)

Capital cost, from Table 2 in Canadian dollars		145,000,000
Revenues at 95,000 tons per year average output	**Canadian dollars per ton of concentrate**	**Canadian dollars per year**
Smelting charge (end 1978 Europe-Japan avgerage) (United States dollars) 85	100.00	
Wage escalation, 0.6 factor	7.20	
Fuel escalation, 0.3 factor	31.15	
Lead price escalation at (United States dollars) 0.45 per pound	0	
Subtotal, smelting charge, end 1980	138.35	
Ocean freight saving (Europe or Japan, average)	32.55	
Refining and marketing charge (Europe or Japan average) 1,254 pound lead payable per ton concentrate at (United States dollars) 0.045	66.39	
Values recovered without payment to mine Lead, 95 percent recovery at 30 kilograms per ton minimum deduction	0	
Silver, 50 grams per ton concentrate at 95 percent recovery	34.04	
Copper, 0.5 percent at 6 percent recovery	4.79	
Sulphuric acid, 0.54 ton per ton concentrate	5.40	
Total revenue, 166,670 tons concentrate per year	281.52	46,900,000
Total operating cost, from Table 7		21,800,000
Depreciation, 5 percent of construction cost, from Table 2		5,900,000
Operating profit		19,200,000
Estimated rate of return, before general/administrative and sales expense and before cost of interest and income tax		13.2 percent

Table 9.
Cost of concentrates (per ton)

	Dollars
Lead, 95 percent recovery at 30 kilograms per ton minimum deduction, 125,416 payable per ton	664.62
Silver, 30 grams payable per ton	21.50
Subtotal	686.12
Smelting charge, from Table 8	138.35
Ocean freight saving, from Table 8	32.55
Refining and marketing charge, from Table 8	66.39
Net payment to mine per ton	448.83

Table 10.

Variation 5 percent +/−	**Approximate effect on return, percent +/−**
Lead price	1.4
Treatment charges	7.0
Energy cost	0.7
Capital cost	1.0
Operating rate	7.3

The authors are grateful for the valuable assistance provided by L. S. Gormely of Wright Engineers Ltd., H. A. Halvorson, and R. I. Burton, independent consulting engineers, all of Vancouver, British Columbia. Likewise, the advice and lead cellroom photograph contributed by Mitsui Mining and Smelting Company are appreciated. ■

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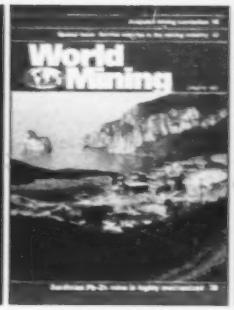

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THE COVER

Concentrator plant at the Masua lead-zinc mine in southwest Sardinia. Samim has recently taken over management of all the base metal mines on the island; Masua is the most modern. It is a highly mechanized mine producing over 20,000 tons of zinc and 9,000 tons of lead each year, the major share of total Italian output. Although Masua is the current showpiece of the Samim group, development and modernization plans exist and are in progress for most of the group's other base metal mines (WORLD MINING, December 1980, page 7). *Photograph by John R. Chadwick.*

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Some 1,500 members of the Association of Mining, Metallurgical, and Geological Engineers of Mexico gathered to view mining equipment and hear 38 technical presentations

Acapulco mining convention and equipment show

By **Donald A. Fazour**
Managing Editor

Every two years, the Mexican mining industry takes a hard look at itself and examines what is new in mining technology and equipment. From October 25 to 29 last year, key mining engineers from this country convened in Acapulco, Guerrero for a technical convention held in coordination with a mining equipment exhibition.

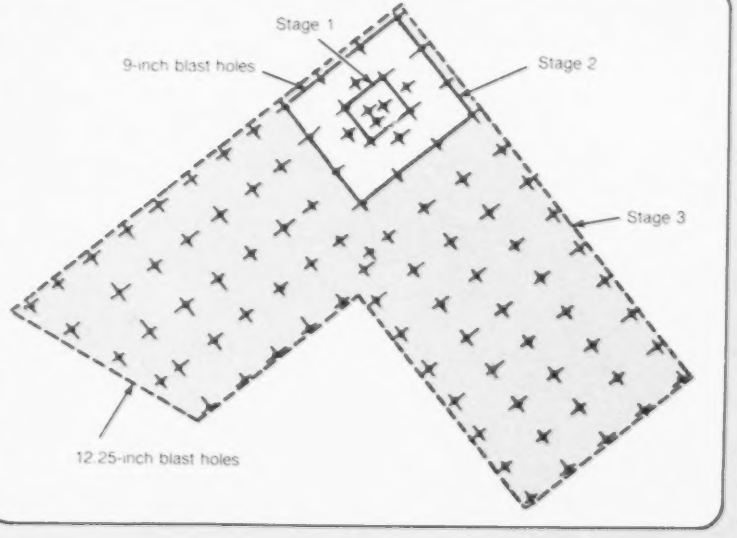
The mining industry of Mexico has long been an important contributor to the country's economy. It has been a growth industry as well. Mining is not growing fast enough, though, according to Ing. Fernando Hiriart, Undersecretary of Mines and Energy. In his opening remarks at the convention inauguration, Hiriart noted that while mining has grown by 48 percent in real terms over the past decade, industry in general has grown by 81 percent. Many old mines are expanding and new mines will start up in the coming years to increase this rate of growth.

While the lowered price of silver has been discouraging to many of the delegates attending the meeting, there was a general sense of optimism. Mexican mines for the most part are high grade. Engineers from the Mexican mines indicated that the emphasis at their operations is for greater mechanization and higher utilization of existing equipment. As the mines mechanize, in addition to allowing the companies to cope with the lower metals prices, lower grade material becomes ore.

Modelling Huge Copper Deposit

Some of the most important mines in Mexico have operated for scores of years. The Cananea mine, currently operated by Compañía Minera de Cananea S. A., has been in operation since 1399 and is still the largest known copper deposit in Mexico. For years, Cananea has been the largest copper mine in the country, but as La Caridad in the same state, Sonora, approaches capacity, Cananea will become the second largest mine—Cananea mines some 35,000 tons of ore per day, and La Caridad will mine on the order of 75,000 tons per day.

Cananea pillar blasting plan **Figure 1**



In one of three papers focusing on Cananea presented at Acapulco, Ramón Ayala and Daniel Chávez reported on a three-dimensional mathematical model of the Cananea ore deposit and described applications of this model. Before any modelling for any deposit can be started, a structured data base is needed. Cananea has, because of its long history, a substantial data base, but a great deal of work was required to structure this.

Geologists at Cananea had to catalog information on 5,183 drill holes, representing 425,000 meters (1,390,000 feet) of coring. Information from these cores was entered on 406,000 computer cards.

Because of the magnitude of the data base, a secondary data base for use with the model was created on the basis of composite information for 10-meter (33-foot) maximum increments of core information. Exact size of each increment was determined on the basis of geological criteria such as rock type, mineralization, etc.

The structure of the model consists of a matrix of equal-sized prisms. Each prism has a 25 by 25-meter (82 by 92-foot) horizontal surface dimension and a 15-meter (49-foot) height. Each prism is coded for rock type, mineralization, copper and molybdenum assays, and the value of variance for use in geostatistical analyses. The size of the modelled deposit is 6,000 by 6,000 meters (19,600 by 19,600 feet) to a depth of 1,200 meters (3,900 feet). The model is used for:

- Determining the distribution and classification of rock types
- Localizing geostructural elements
- Calculating reserves for different cut-off grades
- Geostatistical analyses
- Mine design
- Economic evaluations.

One special considereation in establishing the model is the effect on the deposit resulting from hollow or backfilled underground workings from past operations. A special program to account for partially mined, totally mined, or backfilled workings was developed to address this consideration.

Caving a Pillar for Safety

Quite apart from this mathematical modelling, old workings at Cananea are a major consideration in current open pit operations. This was the topic of a paper presented by José González, mine superintendent at Cananea. Open pit mining in Cananea began in 1944; underground operations, which began in 1899, terminated in 1960. Some of the past underground workings are now encompassed by the planned final pit limits.

González reviewed one particularly difficult safety problem that will be created when the mining level nears one of the larger open stopes, and explained how Cananea was confronting the problem. The stope, called 2-62, is located in the central portion of the "Colorada-Veta" open pit (See WORLD MINING September 1978, pages 64-70). The open space is 400,000 cubic meters in size and would require 680,000 tons of material to fill.

Through review of past data, rock mechanics studies, and careful evaluation of equipment weight and usage, it was determined that until the floor of the pit reached within 30 meters (98 feet) of the roof of this stope, operations in the pit could proceed safely. This dimension is equal to the width of the stope and equal to two benches in the open pit operation.

Once operations reach this level, a three-stage plan to cave the pillar above the stope is envisaged. First, an 8-meter (26-foot) diameter slot will be opened with seven 9-inch diameter drill holes, which will be blasted. The location of this slot is at the vertex of the stope (see Figure 1). The second stage will involve drilling 16 more holes and expanding the corner slot to 18.5 by 18.5 meters (60 by 60 feet). The third and final stage involves blasting the entire pillar and allowing it to cave into the stope. For this stage, a total of 75, 12.25-inch blastholes will be drilled. The shape of the stope and plan for caving it are illustrated in Figure 1.

Recovering Taxco Pillars

Cananea is not the only mine in Mexico faced with difficulties arising from past operations. At the Guerrero mine operated by the Taxco Division of Industrial Minera México S. A., some 416,267 tons of material grading 5.3 ounces silver per ton, 2.56 percent zinc, and 1.58 percent lead has been left in pillars within the mine stopes. This ore can only be recovered by backfilling the stopes, and this could be costly.

Engineers from Industrial Minera México have been studying this problem and seeking an economical way to refill the stopes and recover the ore. The studies that have been carried out were described in a paper presented by Gabriel Zendejas and Carlos Silva. The studies were based on using pneumatically fed dry tailings or sand to refill as mining of the pillars progressed.

Mining in the past has been through shrinkage stoping. The proposed recovery system of remaining ore envisions a 360-meter (1,180-foot) long by 3.9-meter (13-foot) wide operation. A concrete floor would be installed at the base of the proposed operations, through a depth of some 170 meters (560 feet), from level 1 to level 5. Two 1.8-meter (6-foot) diameter raises would be bored about 100 meters (330 feet) from either end of the operation from level 1 to level 5, to be used as ore passes to level 5, where haulage facilities exist. Two existing manways, on the north and south ends of the operation, would be rehabilitated. A 6-foot (1.8-meter) diameter raise would be bored from surface to level 1 for delivery of dry tailings or sands to the pneumatic injector. An 11-inch (280-millimeter) pilot hole would be drilled for an 8-inch (200-millimeter) pneumatic line to feed the fill material into the stope. Haulage within the stope would be by a 2-cubic yard (1.5-cubic meter) captive load-haul-dump unit. The proposed scheme is shown in Figure 2.

A financial analysis of the proposed scheme showed a 57.7 percent profit based on the value of a ton of the ore at current prices. Total cost per ton including mining, milling, indirect costs, and depreciation would be about $23.50.

New Peñoles Properties

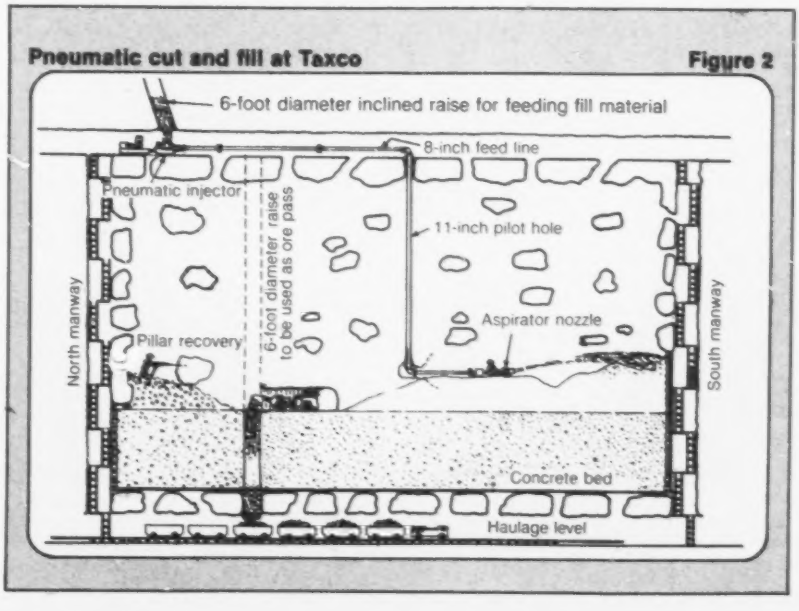
Five separate papers by staff members of Servicios Industriales Peñoles S. A. de C. V. showed how active this company has been in prospecting for future mining operations. The most advanced project described was the La Minita property in southwest Michoacán. Exploration to date on the property comprises 5,960 meters (1,950 feet) of drilling in some 78 exploration holes. Reserves are estimated at 6,400,000 tons of ore grading 2.3 ounces silver, 0.23 percent lead, and 1.27 percent zinc, and 58.79 percent barium carbonate and barium sulphate per ton.

The proposed operation for which feasibility studies are complete would include a 2,000-ton-per-day open pit operation with a 1.68 stripping ratio and a 52° pit wall slope. Laboratory and pilot plant metallurgical testing has been completed and a preliminary design for a 2,000-ton-per-day mill completed. Two papers on this operation were presented: one on plant design by Carlos Gómez and Fernando Grajales; one on pit optimization by Victor de la Garza, José Gaytán, Fernando Grajales, and José Olmos.

Another important Peñoles discovery is the Gohico silver-zinc-lead deposit some 30 miles (48 kilometers) north of Alamos in the northern state of Sonora. This deposit was dealt with in two presentations as well: the geology of the deposit was discussed by Luis Calientes; mine planning by Ignacio Martínez and Fernando Grajales.

Gohico has proven and probable reserves of 801,000 tons grading 4.7 ounces silver, 9.22 percent zinc, and 0.45

Pneumatic cut and fill at Taxco **Figure 2**



percent lead per ton. A 500-ton-per-day underground operation is planned with a projected start-up in April of this year. The mine will have a life of some five years at a mining rate of 500 tons per day. Mining and milling costs are expected to be $23.50 per ton, including direct and indirect costs. Even at today's low prices, Gohico should be a very profitable operation for Peñoles.

The final Peñoles property discussed by Javier García, Luis Novelo, and Juan Pérez, was El Rey de la Plata, a property that is undergoing extensive evaluation by Peñoles, but one which is not as far along in development as Gohico and La Minita. The paper that they presented discussed the general geology of the deposit and district where it is located, in the north-central part of the state of Guerrero.

The mine would consist of two separate deposits, El Rey de la Plata itself and the Tehuixtla deposit. Total estimated reserves to date are 1,700,000 tons grading 8.8 ounces silver, 1.7 percent lead, and 7.8 percent zinc per ton. There is still much exploration to be done on this property to verify and test for further reserves, the authors reported.

Huge Pipeline Project

One of the most interesting papers at the convention was presented by Raúl Morales of Altos Hornos de México S. A. The paper described one of the largest slurry pipeline projects in the world, the 239-mile (384-kilometer) iron ore concentrate line that will carry concentrates for pelletizing from the La Perla and Hercules mines to the plant in Montclova through the states of Chihuahua and Coahuila. Table 1 shows the relative size of the new project to the other large slurry pipelines in the world.

Total cost of the pipeline will be $76,000,000 and the line should be operational late this year. Slurried iron ore will be transported at 60 to 70 percent solids through a two-stage system. The first stage begins with two 1,250-horsepower pumps at La Perla. The slurried concentrate will be pumped to Hercules, 55 miles (89 kilometers) away, at a rate of 202 tons per hour through an 8-inch (200-millimeter) diameter pipeline.

La Perla concentrate will be mixed with slurried concentrate from Hercules at this point. The combined concentrates will be pumped using three electric 1,500 horsepower pumps at a rate of 606 tons per hour through a 14-inch (355-millimeter) diameter pipeline to a third and final pumping station in the desert. The line from Hercules feeds directly into the pumping station. The desert pumping station comprises three 1,500 horsepower Diesel-powered pumps. Distance from Hercules to the pelletizing plant in Montclova is 184 miles (296 kilometers).

The above reviewed papers are but a few of many excellent and practical presentations heard by the Acapulco delegates. A large number of papers were concerned with geological studies conducted throughout Mexico. One paper described one of the first uranium occurrences being examined in the state of Sonora in northern Mexico. There were excellent mineral processing papers as well. Metallurgists from Industrial Minera México reported on research conducted on lead-copper differential flotation. Engineers from Compañía Real del Monte y Pachuca S. A. described tests of theirs that used lamella thickeners for counter-current decantation of silver-gold leach slurry. All papers were made available to the delegates in a 1,445-page hardbound proceedings volume.

Table 1
Largest slurry pipelines of the world

Project	Miles	Metric tons per year	Pipe diameter (inches)	Start up	Material slurried
Black Mesa Arizona, United States	275	5,000,000	18	1968	coal
Samarco Brazil	248	12,000,000	20	1977	iron ore
La Perla Mexico	239	4,500,000	8/14	1982	iron ore

FIRST TO VIEW the equipment exhibit in Acapulco were, left to right, Ing. Fernando Hiriart, undersecretary of Mines and Energy; Alejandro Cervantes Delgado, governor of the state of Guerrero; and Ing. Ricardo Esquivel Esparsa, head of the organizing committee.

Well-Paced Program

Technical sessions were scheduled from 10:00 a.m. to 2:00 p.m. This allowed delegates ample time for personal discussions and for viewing the equipment exhibited at the Acapulco convention center. Most all equipment manufacturers from Mexico, the United States, and Europe were present at the show, and engineers had the opportunity to see most of the 15,000-square meter (161,500-square foot) exhibit.

Manufacturers, who can frequently gauge potential growth of an industry they serve, were enthusiastic about Mexico. As important to the growth of the industry as new mines are those mines that are being expanded. The newsletter on pages 7 and 9 of this issue clearly documents the magnitude of the expansions underway in Mexico.

Because it is a government policy to favor domestically produced products over those imported from abroad, more and more companies are either in the process of or are considering establishing manufacturing facilities in Mexico.

The Acapulco show has been growing each year and the next convention, to be held in late fall, 1983, should be larger than ever. Although specifically aimed at Mexican mining engineers, it should be well worth attending by non-Mexican mining professionals as well. ■

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World Mining

FEBRUARY 1982

INTERNATIONAL GUIDE TO BLASTHOLE DRILLS

FEBRUARY 1982
VOLUME 35 NO. 2

World Mining

THE COVER

Blasthole drilling in a northern open pit mine. There are many trends in blasthole drilling today, such as increased demand for angle drilling, new markets for electric drills, and a keener interest in long bit life. John Chadwick has assembled the facts in a special report.

Features

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Ready reference specifications, photographs, and an analysis of blasthole drilling today are all in this comprehensive report by John Chadwick

Drilling for Profit 40
A companion piece to the International Guide, this feature tells you how to obtain maximum production at minimum cost through good management. Roy Moore of Reed Mining Tools, Inc. is the author

Borehole Surveying 42
Increased efficiency in exploration and mining is obtained by accurate surveying. Randy Cobbley of Eastman Whipstock, Inc. describes the latest technology

Cobalt Today and Tomorrow 48
A generally optimistic view prevailed at a recent international cobalt conference, held in Brussels, Belgium. John Chadwick was there, and has filed this report

World Mining Congress 53
The 11th edition of this uniquely important convention will be held in Belgrade, Yugoslavia in just a few short months. George Argall, who is on the Congress' Organizing Committee, has all the details in a pre-Congress outline

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The mood was one of cautious optimism. As ever, when cobalt is discussed, Zaire and Zambia were the focus of attention. There was great diversity of opinion concerning the potential of cobalt production from undersea nodules

Cobalt today—and its future recovery and use

By **John R. Chadwick**
Associate Editor

Despite the weak cobalt market that has prevailed for almost the last two years, the "International Conference on Cobalt—Metallurgy and Uses" held in Brussels, Belgium, in mid-November 1981 attracted some 300 delegates from almost 30 countries. The conference brought together producers and users and was significant for the generally optimistic view of the future of the cobalt market held by a large number of those attending. This optimism carried over, in some people's minds, to the copper market.

Appropriately, the first session was opened by a paper from the leading producer country, Zaire. This session was devoted to cobalt resources and the questions of supply and demand. Lukusa Muengula, of Sozacom, presented "The Cobalt Future in the Competitive Strategy." Muengula predicted that for the next five years, Zairean production of cobalt would rise at 4 percent per year; production in 1980 was 14,500,000 metric tons of metal. This increased production is to come from previously undeveloped reserves and the redevelopment of old mining areas.

Muengula quoted a U.S. Bureau of Mines figure for an expected 3.5 percent growth rate in United States cobalt consumption. It was later pointed out by one of the delegates that the U.S. Bureau of Mines had subsequently stated that permanent substitution of cobalt in the United States now amounted to a figure of almost 910,000 kilograms (2,000,000 pounds). In addition, Muengula quoted a growth rate of 3.3 percent for cobalt consumption in the rest of the western world.

He highlighted the need for stabilizing the producer price and urged for better cooperation between cobalt users and producers. On this point, he informed the conference that on the previous day, November 9, 1981, the cobalt producers, in a meeting in Brussels, had agreed to set up a center for the development of cobalt. (see WORLD MINING, January 1981, page 67). The aims of the center would include promoting the use of the metal and providing technical assistance to the end users.

The second paper, entitled "Cobalt Resources: Reserves and Availability," was presented by P. Scheidweiler of Société Générale des Minerais. He began by discussing the current supply situation; some 95 percent of western world cobalt production comes, as a by-product to copper production, from Zaire and Zambia, or as a by-product from nickel mining elsewhere in the world; the world's only primary cobalt producing mine is the Bou Azzer in Morocco (see WORLD MINING, March 1980, page 7).

The known, mineable reserves of cobalt in the world today amount to between 3,000,000 and 5,000,000 tons. In addition, there are a further 4,000,000 to 5,000,000 tons classified as potential reserves. Interestingly, the largest tonnage is to be found in Cuba, 800,000 tons, but this cobalt is in lateritic deposits and cannot be extracted economically as yet. Similarly, large reserves in Indonesia are also uneconomic to mine at present. Zaire, the number one producer, has reserves of 450,000 tons.

At present, less than 25 percent of the mineable reserves are being exploited; another 25 percent of these reserves are under study; and over 50 percent of the mineable reserves are receiving no attention whatsoever. At present, of the 800,000 tons of mineable reserves that are being mined, 70 percent are in central Africa. Of the 900,000 tons of mineable reserves that are under study, only 40 percent are in central Africa. Further, pointing to even less dependence on this area of the world, of the remaining 2,200,000 tons of mineable reserves, only 15 percent are in central Africa.

However, breaking the mineable reserves up by quality presents another interesting picture. High grade reserves amount to 400,000 tons and are all in central Africa; medium grade reserves total 950,000 tons, of which 47 percent are in central Africa; finally, the largest tonnage, 2,550,000 tons, is in low grade reserves, of which only 15 percent are to be found in central Africa. Dividing the total mineable reserves by annual production gives a reserves life of 80 years. Dividing high grade reserves gives a life of just 27 years.

Scheidweiler then went on to discuss the implications of undersea mining. Identified cobalt resources undersea amount to 500,000,000 tons, while on land they amount to just 10,000,000

Table 1.
Substitution and growth potential for cobalt by use

Use	Substitution Potential Short Run	Substitution Potential Long Run	Growth Potential
Superalloys	Minimal	Moderate	Moderate; aerospace slowdown, new fabrication methods
Magnetic Alloys	Major	Major	Low, unless markets develop for cobalt-rare earth magnets
Carbides and or Hard Metals	Minimal	Minimal	Moderate; strong energy-related demands partially offset by recycling and new products
Tool Steels	Moderate	Moderate	Low
Hardfacing	Moderate	Moderate	Low
Catalysts	Minimal	Minimal	Good; may be offset by recycling
Other Chemicals	Moderate	Moderate	Moderate

tons. However, many factors still militate against seabed mining, including the vast investment sums that will be required, and the fact that the Law of the Sea has still to be agreed upon. There has been no significant production from this source to date. However, there will come a time, when the profitable reserves have been exhausted, when the choice has to be made between developing the marginal land reserves or commencing seabed mining.

In discussing the cobalt pipeline, Scheidweiler pointed out that to achieve security of supply, a working pipeline should contain about 20,000 tons of cobalt, the pipeline being the amount of cobalt in transit, in various situations, between the producer and the user. He went on to discuss stockpiles and differentiated between the United States stockpile, which is truly strategic, with military requirements taking first priority, and the Japanese and European stockpiles, which are established mainly for economic reasons. He pointed out that the current level of the GSA stockpile is about 20,000 tons but that the goal for the future is to raise this level to 37,000 tons. In conclusion, Scheidweiler stated, "cobalt still looks promising."

J.C. Agarwal, of Charles River Associates, presented "Role of Cobalt in the Industrial Society." He began by noting the 1980 world consumption of 22,000 tons. He went on to consider cobalt's uses, and reported that its high temperature characteristics were still the most important. No replacement is likely to be found for cobalt in the near future for such applications. In fact, when considering substitution in such uses, it is not price that is the determining factor, but availability. Oil industry usage in catalysts is also still on the increase, particularly in connection with sour gases and oil. Table 1 shows the predictions for substitution and growth potential of cobalt by use.

The ratio of the cobalt price to the copper price, which was 8 to 1 at one time, has now risen to 25 to 1, a great incentive to increase cobalt recoveries. On the subject of seabed nodules, Agarwal reported that the cobalt grade of these deposits ranged from 0.1 to 0.25 percent. It is very difficult to recover cobalt from this source, he said. In fact, laboratory test work has only achieved recoveries of around 30 percent and in his opinion, 50-percent recoveries would be very optimistic. A 3,000,000-ton per year manganese plant based on seabed nodules could produce 3,600 tons of cobalt per year, but it would cost some $1,500,000,000 to set up with a further $250,000,000 to develop the technology. In his opinion, seabed mining is still 10 to 20 years away, and the latter is most probable.

However, Agarwal also concluded on a note of optimism, saying, "as long as the cobalt supply is assured, people will find new uses for the metal."

Cobalt Optimism

The final paper covering the overall cobalt market was, "Cobalt—a Critical Material?" by Jorn P. deLinde, a report on the cobalt aspect of an extensive research program into critical ferroalloys carried out by Chase Econometrics.

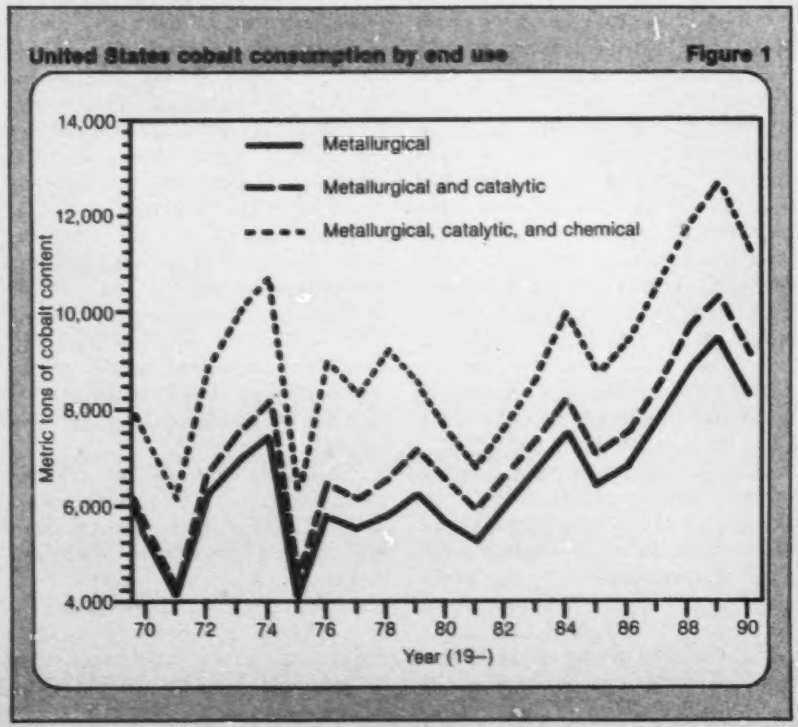
The year 1979 was not a good one for cobalt in the United States. Total consumption declined by more than 7 percent. Primary consumption fell even more as secondary recovery continued to advance. This malaise was caused by the tight supply situation and consequent sharp rise in price, spurring considerable substitution and conservation efforts by cobalt users. Preliminary information for 1980 indicated an 11 percent drop in total United States consumption. Figure 1 shows the prediction for future cobalt consumption in the United States. Peak consumption occurs in 1989, at almost 13,000 tons.

In discussing substitution and whether it is permanent or whether users will return to cobalt at lower prices, deLinde made the following, very interesting point. "There may be a fundamental difference between civilian and military markets in this respect. In civilian applications price, cost, and performance are always part of a trade-off, whereas the military generally wants the highest performance that is technically possible regardless of cost." The Chase forecast does assume that there will be "a fair degree of reversible substitution." Looking to total western world consumption, stronger advances are expected outside of the United States and Japan in the coming decade.

Considering the security of supply, there is no absolute shortage of reserves. The problem is that vast quantities of the reserves are situated in countries facing political or economic difficulties. About 2,000,000 tons of cobalt were classified as reserves by the U.S. Bureau of Mines in the 1980 edition of Mineral Facts and Problems. Of this total, almost 78 percent is located in Zaire and Zambia. Reserves outside of these countries are only the equivalent of 16 years demand at the 1974 level, while reserves including Zaire and Zambia total 73 years demand at this level. There are no significant cobalt deposits classified as reserves in the major industrialized countries at present.

When considering the fact that the majority of cobalt production is a by-product of copper production, deLinde made the following observation. "We view copper as seriously underpriced rel-

United States cobalt consumption by end use — Figure 1



ative to long-term costs as we enter the 1980's, and the lack of adequate growth in copper mine capacity is likely to produce a much tighter copper market by the mid-1980's than we are currently experiencing. The consequent sharp rise anticipated in copper prices serves to diminish production costs for cobalt through the mechanism of by-product credits."

The conclusion of the Chase Econometrics research is one of optimism for the cobalt user: "The underlying economics of cobalt supply in the 1980's look relatively favorable, particularly with regard to the expected increase in the profitability of associated copper mining."

Cobalt Extraction

Professor J. De Cuyper of the University of Louvain began the session on cobalt metallurgy by reviewing "Concentration of Cobalt Ores." He commenced with a look at the main cobalt ores being processed: cobalt in the form of sulphides, arsenides, sulpho-arsenides,and oxidized cobalt minerals. De Cuyper reviewed the various physical concentration techniques available for cobalt ores and gave details of the flowsheets of many of the cobalt concentrators around the world, including the Gecamines' plants in Zaire; the Zambian concentrators at Baluba, Rokana, and Chibuluma; the concentrator at the Bou Azzer mine; the Kilembe plant in Uganda; and concentrators in Finland and Japan. At the Bou Azzer concentrator, for instance, gravity methods are used on the cobalt arsenide ore to produce concentrate with a Co content ranging from 6.5 to 17.5 percent.

Concerning laterite ores and seabed nodules, De Cuyper does not see much potential for physical concentration techniques. Looking to the future, improvements in cobalt concentration are foreseen for the operations in Zaire and Morocco, through greater selectivity. Selective chelating agents, dealt with in a later paper, have great potential for improving future recoveries. The emphasis, up to now, has not been on improving cobalt recovery. This situation is likely to change in the future.

Reijo Anttonen presented, "How Outokumpu Beneficiates the Low Cobalt Contents of its Sulphide Copper Ores." Cobalt concentrates are produced by flotation at the Keretti and Vuonos mines. These cobaltiferous pyrite concentrates are further treated at the Kokkola cobalt plant. The head grades of the Keretti and Vuonos ores are 2.7 and 1.9 percent Cu, 0.6 and 1.1 percent Zn, and 0.23 and 0.12 percent Co, respectively. The cobalt concentrate from Keretti contains 0.7 percent Co, and from Vuonos 0.9 percent Co.

The cobalt carriers of the Keretti ore are pyrite (0.8 percent Co), cobalt-nickelpentlandite (33 percent Co), and pyrrhotite (0.1 percent Co). The Vuonos ore is geologically similar but there are only minor amounts of pyrite. In both ores the cobalt-nickelpentlandite occurs partly as extremely fine ex-solutions in pyrrhotite. Selective flotation is the method used, with on-stream analysis. A pilot cobalt flotation plant has been tested, but not with much success as yet. At Vuonos, cobalt scavenging has been introduced successfully. The recovery at this concentrator runs at 51 percent Co. Anttonen also discussed the influence of the degree of grind; trials of different reagents; the effect of process water circulation; and economic optimization of copper-cobalt separation.

Moving further along the cobalt production process, Doctor Claude Feneau, of Metallurgie Hoboken-Overpelt, discussed "Extractive Metallurgy of Cobalt." This was a review of many of the most important cobalt plants in the world and the various routes available for the production of finished cobalt.

Processes based on primary cobalt concentrate include roasting, leaching, and electro-winning in sulphate media. In the case of arsenical ores, leaching and electro-winning are carried out in chloride media. Secondary feeds, such as mixed cobalt-nickel sulphides, cobalt-nickel mattes, or alloys, are processed in different media, including sulphate, chloride or ammoniacal. Feneau also considered very specific processes for the treatment of slags, nodules, and spent catalysts, and commented that recycling of the latter will increase.

"Sumitomo's Cobalt Refining Process" was described by N. Ono. Sumitomo Metal Mining Co. Ltd. (SMM) operates a hydrometallurgical plant for nickel and cobalt recovery from their sulphide mixtures and hydroxide precipitates produced at the Niihama nickel refinery. The outline of the process is as follows:

1. Pressure leaching of the sulphide mixtures
2. Purification of the leached solution
3. Nickel and cobalt separation by solvent extraction
4. Gypsum production and ammonia recovery from ammonium sulphate solution from the solvent extraction process
5. Production of electrolytic nickel and cobalt from their respective chloride solutions by electro-winning
6. Recovery of chlorine gas from electrowinning and the production of hydrochloric acid

The plant produces 2,500 tons of electrolytic nickel and 1,500 tons of electrolytic cobalt, at full capacity, with some by-products such as nickel chloride, cobalt oxide, cobalt nitrate, and cobalt sulphate. The finished products contain 99.98 percent nickel and 99.94 percent cobalt, respectively. The major impurities in the electrolytic cobalt are 0.004 percent Cu, 0.005 percent Fe, 0.003 percent Zn, and 0.001 percent S.

Considering the world's major producer, Tshofu Shungu, of Gecamines, described "Recent Improvements in Metallurgical Processing at the Cobalt Granules Plant of Shituru." Here, improvement of the quality of the cobalt product has depended on improving the conditions of electrolysis and thermal refining. In electrolysis, Gecamines conducted research into the effect of pH, the influence of current density, and the effect of temperature. In the laboratory it was found that, the current efficiency initially increases with pH up to a value of 6.2. Then, however, it decreases with increasing pH. A higher pH produces a considerable increase in nickel and sulphur contamination.

Increasing current density results in a decrease of current efficiency. At a current density of about 220 amps per square meter (and a current of about 7,000 amps), the current efficiency is over 90 percent. The efficiency falls to about 80 percent when the current density is increased to 375 amps per square meter.

In the laboratory, higher temperatures of electrolysis result in improved current efficiencies and lower nickel and sulphur contamination. On moving to industrial scale tests, Gecamines had to change to sandblasted stainless steel blanks to successfully achieve the benefit of higher temperature electrolysis. Table 2 shows the improvement that was achieved by electrolysis at 80° C, compared with electrolysis at 60° C.

Gecamines examined three improvements: the quality of the reactives, the method of charging the furnace, and operation control, with a view to increasing the efficiency of thermal refining. The first step to control iron contamination by the reactives was to replace sand by CaF_2. This helped to increase both the fluidity of the slag and its desulphurizing capability. Also, silico-calcium was replaced by a better sorted and less oxi-

dized product (silalcium). This allowed the sulphur content of this reactive to be reduced from 1,276 to 1,103 parts per million.

It was also found to be beneficial to replace rock lime with crushed lime and to inject it directly into the furnace bath, thereby further reducing iron contamination. With a view to improving sulphur contamination control, Gecamines will buy equipment to rapidly analyze sulphur contents.

The concentrates produced at the Bou Azzer mine in Morocco are sent to Savoie in France for further processing. This plant was described by F. Peyron in "A French Producer of Cobalt: Metaux Speciaux S.A." The concentrate received is first treated by roasting, to recover the arsenic as a high purity trioxide. Subsequent hydrometallurgical treatment of the calcine is as follows:

- Hydrochloric leaching in the presence of gaseous chlorine
- Recovery of dissolved gold
- Solution refining to recover Cu, Bi, Ag, Zn, Cd, Pb, Ni, and Fe
- Electrolytic production of metal from cobalt chloride
- Refining of spent electrolyte, yielding cobalt oxide, which is the basic source of by-products such as cobalt salts and oxides.

Although this process was specifically developed for the hydrometallurgical treatment of cobalt arsenides, it can be adapted to the treatment of sulpho-arsenides and other cobalt-bearing raw materials. For the electro-winning of the cobalt, cells and anode compartments made of plastic are used. The anode is made of graphite and the cathode of stainless steel. The current efficiency is around 90 percent, at a temperature of 60° C and with a cathode density of 350 amps per square meter.

"The Recovery of Cobalt from Laterite Ore" was discussed jointly by L.A. Goeller and C.D. Low of UOP Inc. Nickel laterite ores are a large and still largely undeveloped source of both cobalt and nickel, representing some 65 percent of cobalt reserves. Reductive roast-ammonia leach technology offers good recovery of both cobalt and nickel—70 percent of the cobalt and 90 percent of the nickel. The cost of the cobalt production is estimated to be in the neighborhood of $4.50 per kilogram ($2.00 per pound).

UOP's Nickel Extraction Process relies on the use of additives in the roast step to increase metal recovery from a wide range of ore types, including the more refractory saprolitic ores. The additives include various sulphur forms (gaseous and solid) and halides. Figure 2 shows the roast and leach sections of the UOP Process. The process and other alternatives were described in detail, including a good deal of information on operating and capital costs.

M. Bourahla, of Sonarem, Algeria, described laboratory work that had been carried out for the "Purification of Cobalt Chloride Solutions—The Winning of Very Pure Metal and its Characterization." Iron was removed from a chloride solution using solvent extraction, with M.I.B.C. Then ion exchange in a Dowex resin column was carried out to remove the nickel. Low intensity electrolysis was carried out to eliminate elements such as copper and lead, and then the main electrolysis was carried out to produce very pure cobalt. Ultimate purification was achieved by zone melting or thermal treatment.

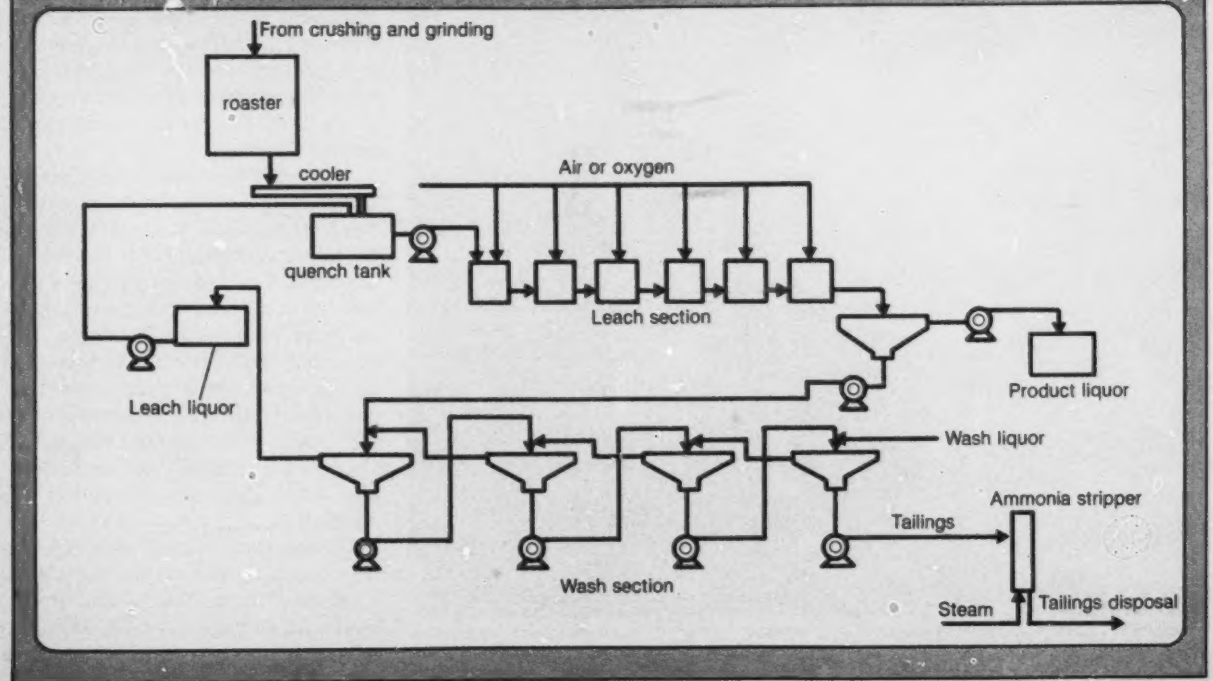
In "The Chemistry and Thermodynamics of a Hydrometallurgical Sulphite Route for the Recovery of Cobalt," D.C. McDonald, P.B. Linkson, and

Table 2.
Quality of cobalt obtained by electrolysis and from reference cell

Impurity content (parts per million)	Co	Cu	S	Fe	Ni	Pb	Zn	Ca
Electrolysis at 60° C	98.9 percent	75	650	70	960	40	8,600	107
Reference cell at 80° C	95.8 percent	58	1,100	275	1,280	110	22,000	321

1. With exception of Co.

Roast and leach section, UOP process **Figure 2**



D.M. Nobbs, of the University of Sydney, Australia, discussed the recovery of cobalt sulphites from aqueous solutions by addition of sulphur dioxide and subsequent adjustment of pH. The effects of temperature and of the spectator ions, sulphate, nitrate, and chloride on precipitation of the cobalt sulphite were elucidated by a thermodynamic model for metal sulphite precipitation, with experimental determination of species in the solutions. The recovery of cobalt from solutions containing nickel and/or copper via routes involving sulphite salts was also covered.

D. Pearson of Warren Spring Laboratory, United Kingdom, closed the metallurgy session with "Recovery of Cobalt from Metallurgical Wastes." Some of the processes to recover cobalt from sludges and grindings were reviewed and a new laboratory process was described. In this, major metal values are dissolved in sulphuric acid and separated by precipitation and solvent extraction, using a newly-developed extractant. High purity metal is then electro-won. Data presented showed that costs range from $4,500 to $7,300 per ton of cobalt recovered.

A further 16 papers were presented on the uses of cobalt. There were also a series of poster sessions, with some papers of great interest to the cobalt producer. G. Rinelli and A.M. Marabini, from the CNR Laboratory, Italy, discussed the "Possibility of using Chelating Agents in the Flotation of Cobalt Minerals." The presentation included proposed applications for the concentration of refractory ores, as cobalt oxides and/or arsenides.

The "Recovery of Cobalt from Ocean Nodules and its Impact on the World Market" was covered by J.C. Agarwal, J.H.R. Black, and B.J. Reddy, of Charles River Associates. One ocean mining project achieving 80 percent cobalt yield could produce 6,000 tons of cobalt annually. This sort of operation could become important in the 1990's, provided that legal and technical problems are resolved. Five major consortia are involved in this research. If they were all to come on stream in the next decade, there could be 30,000 tons of cobalt for the market to absorb from this source. This would obviously drive the price down.

"A New Generation of American Cobalt—The Anschutz Madison Project" was described by Spisak of the Anschutz Mining Corp. To be on stream in early 1984, this project, centered in the old Missouri Lead Belt, will produce more than 9,000 tons of cobalt and 10,000 tons of nickel during its minimum 10-year life. A bulk sulphide beneficiation circuit is expected to achieve a cobalt recovery of 90 to 93 percent. Copper will also be produced at an annual rate of 15,000 tons.

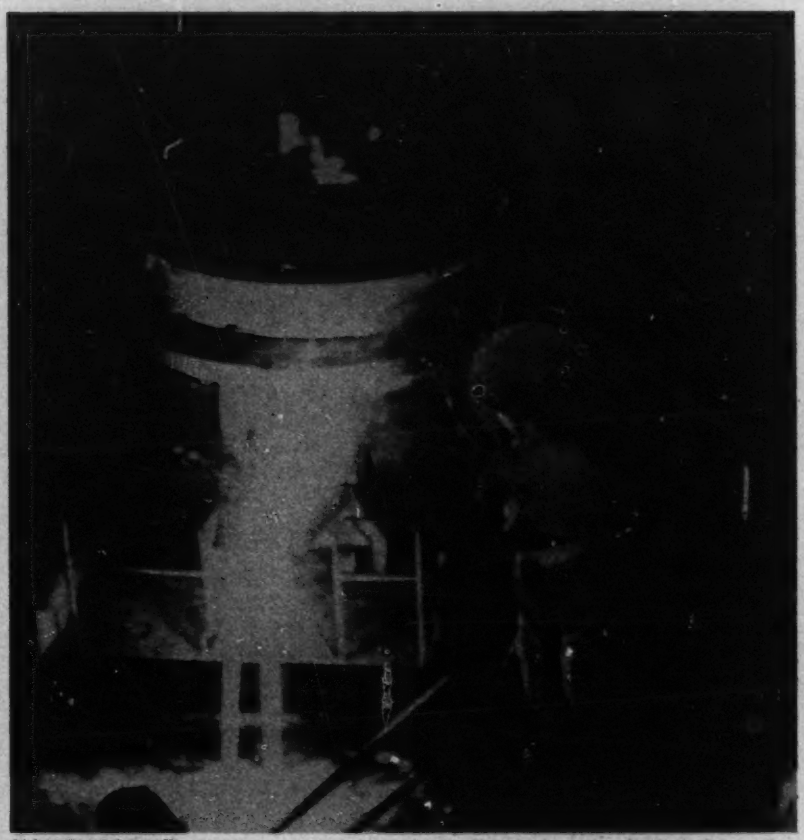
Cyanex CNX was described and its application considered in "Phosphine Chemicals—A New Era in Selective Extractants," by D.R. Burley of American Cyanamid. R.G. Reddy and G.W. Healy, of the United States, reported on "Studies on Removal of Cobalt from Liquid Copper Scrap under Fire Refining Conditions."

Members of the National Metallurgical Laboratory, India, covered "Recovery of Cobalt from Lateritic Nickel-Cobalt Ores with Special Reference to Sukinda Ultramafic Field." The nickeliferous laterite of this area is the only known potential source of nickel and cobalt in India. Reserves of approximately 98,000,000 tons of ore containing 0.8 percent Ni and 0.06 percent Co have been estimated around the Kansa area. Moreover, overburden removed in several major chrome mines in the area has been found to be lateritic in nature, containing 0.5 to 1.2 percent nickel and around 300 parts of cobalt per million.

A study of the Kamoto, Zaire, ores was presented by C. Ek, J. Frenay, and B. Lorea of the University of Liège, Belgium, in "Treatment of an Oxide Copper-Cobalt Ore." The oxide minerals, such as tenorite, malachite, pseudomalachite, and heterogenite, assay 2.2 percent Cu and 3.8 percent Co. Flotation tests were very disappointing, but leaching with sulphuric acid achieved recoveries up to 96 percent for copper and 90 percent for cobalt.

"Cobalt Extraction from Potential U.S. Resources Today and Tomorrow" was presented by B. Yarar of the Colorado School of Mines and L.J. Miller of Texasgulf. There are indications of a cobalt-bearing Permian shoreline extending from Texas to North Dakota. Further, shale beds in South Dakota show similar formation to deep sea nodules and strongly indicate the presence of cobalt along with 25-percent manganese. Other deposits as well as deep sea nodules and metallurgical wastes were also discussed.

The complete proceedings of this conference are available from the Centre d'Information des Metaux Non Ferreux, Boulevard de Berlaimont 12, 1000 Brusells, Belgium. ■



MOLTEN COBALT metal is tapped out of an electric furnace into a water bath, for cooling and granulation. More than half of the cobalt produced in the world is melted this way.

After THOUGHTS

With Southwestern United States copper mines reducing capacity or halting operations, everyone is wondering about the future of one of the most important nonferrous metals markets in the world. The price of copper has been volatile for years and few industry analysts are willing to make long term projections. A New York, United States financial institution, Chemical Bank, has recently taken exception to this general rule and made some very interesting projections for the future of the copper market between now and the end of the century.

According to Chemical Bank, real copper prices will rise about 1.0 percent per year for the remainder of this century, substantially less than generally expected. The principal factor in this forecast is that a satisfactory supply-demand situation is expected. Table 1 shows just what copper price projections have been made by the bank.

Chemical Bank points out in its report that major trends to note in the industry are the growing importance of coproducts in determining mine profitability, the entry of energy-related companies into the market, and the developing oligopolistic control of copper production by major multinational companies and Third World host governments.

Declining demand for copper from traditional users, such as the electrical, telecommunications, and plumbing industries, will be compensated by the development of major new markets for copper such as solar energy, electric cars, and desalination programs. The demand for copper is forecast to rise between 2.0 and 3.0 percent per year over the next twenty years—from about 7,300,000 tons in 1980 to about 12,000,000 tons in 2000.

Capacity is expected to increase in line with the projected growth of world demand. Significant additional capacity is already in place and could come on stream quickly, according to the report. In addition, many recently discovered high grade deposits appear profitable even at current copper prices, while the energy companies are likely to proceed with more rapid development of existing ore bodies.

Overall capital requirements of the copper industry in the period 1982-2000 do not appear excessive. However, some less developed economies may experience difficulties. It is estimated that developing countries' external financing requirements for expansion of the copper industry expansion will aggregate over $1,000,000,000 per year in 1980 dollars through the forecast period. Table 2 shows projected capital expenditure for new capacity through the year 2000, both for cases in which coproducts are taken into account and for cases in which they are not.

Successful projects will be those resilient to cyclical movements in underlying demand. The financial future of most of the independent producers, and those developing countries with marginal deposits, must remain in doubt. Such developments may encourage some developing economies to concede a more direct role to major independent mining companies in the mining of their copper and other mineral resources.

Table 1.
Copper price projections 1980-2000

	Average Price (current dollars)	95% Confidence Interval[1]	Average Price (constant 1980 dollars)
1975-1980	75	—	95.0
1981-1985	98	± 5%	79.4
1986-1990	149	± 9%	83.9
1991-1995	228	± 14%	87.4
1996-2000	360	± 20%	98.7

1. The 95 percent confidence interval indicates a 95 percent certainty that the actual price will lie within the indicated percentage either side of the mean forecasts.

Table 2.
Investment requirements for additions to copper capacity 1980-2000

	Required additions (000 tons)	Total cost (A)	Total cost (B)	Cost per year (A)	Cost per year (B)
		(millions of 1980 dollars)			
1980-85					
Mine	670	2,010	3,200	402	640
Smelter	920	1,640	2,050	328	410
Refinery	960	415	435	83	87
Total		4,065	5,685	813	1,137
1986-2000					
Mine	5,450	16,365	25,900	1,091	1,727
Smelter	6,240	11,700	14,870	780	991
Refinery	6,610	2,800	2,940	187	196
Total		30,865	43,710	2,058	2,914

(A) Includes coproducts valued as copper.
(B) Excludes coproducts

Producers at operating mines have been successful over the past several years in bringing mining costs under control. In the year following October 1973, when the first oil crisis was at its height, average net mine operating costs rose by more than 40 percent. In 1980, by contrast, average costs increased by only 8 percent, notwithstanding escalating oil and energy costs and high general inflation rates in many producing regions.

It is interesting to look at some of the new copper projects in planning or development stages in view of this report. In the case of Ok Tedi, in Papua New Guinea, a project approved and underway, coproduction of gold figures prominently. When looking at energy company involvement, Disputada's Los Bronces property in Chile comes to mind. Involvement of multinational companies with third world host governments is reflected in new mining codes in Argentina, Peru, and Chile, wherein foreign investment is encouraged. High grade deposits that come to mind inlcude the newly-discovered La Escondida deposit of Getty Oil and Utah International. Marginal deposits would include Cerro Colorado in Panama, Quellaveco in Peru, and many of the undeveloped properties in Chile.

The full report on which this column is based is available from Chemical Bank, 277 Park Avenue, New York, New York 10172, United States. Contact Gerald Pollio. —*DAP*